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DEPARTMENT OF COMMERCE

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE OLD DOMINION CO., GLOBE, ARIZ.

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D. L. FORRESTER AND W. B. CRAMER

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MILLING METHODS AND COSTS AT THE CONCENTRATOR OF THE OLD DOMINION CO., GLOBE, ARIZ.[1]

By D. L. Forrester[2] and W. B. Cramer[3]

INTRODUCTION

This paper, describing the milling practice of the Old Dominion Co., is one of a series being prepared by the United States Bureau of Mines.

ACKNOWLEDGMENT

In the preparation of this paper, acknowledgment is made of the valuable service rendered by R. S. Allen, metallurgist of the Old Dominion Co. T. G. Chapman, one of the consulting engineers of the Bureau of Mines, assisted in arranging the paper for publication.

GENERAL DESCRIPTION

The crushing plant and concentrator of the Old Dominion Co. are adjacent to the main hoisting shaft of the Old Dominion mine, which is just outside the city limits of Globe, Gila County, Ariz.

Mining of the ore, along a vein of varying width for a distance of approximately 2 miles, is done by various methods. Square-setting is employed in the lower oxidized ore bodies, caving methods are used in some sections of the concentrating ore, and top slicing and shrinkage in other sections. The caving system of mining is used whenever possible.

The crushing plant consists of the usual equipment in vogue at the time it was built, about 1914, with a few additions which are described in detail under "Breaking."

The concentrator was originally designed for combined gravity and flotation treatment and for a capacity of about 800 tons per 24 hours. At present the treatment is entirely by flotation methods, and the capacity is about 1,500 tons per day.

All of the water used for milling purposes is pumped from the mine, and none of the water used is reclaimed.

The company generates its own power by Deisel engines and an auxiliary steam plant. A new 2,000-hp. Deisel engine was recently installed which will probably eliminate the steam plant entirely, except in emergencies.

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1 The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6467."

2 One of the consulting engineers, U. S. Bureau of Mines, and Superintendent of reduction, Old Dominion Co.

3 One of the consulting engineers, U. S. Bureau of Mines, and Consulting engineer, concentrating department, Phelps Dodge Corporation.

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ORE TREATED

Copper ores, principally chalcopyrite, containing some gold and silver are treated. The term "ores" is used for the reason that there is quite a difference in the character of the gangue from the different ore shoots in the mine. The gangue may be fairly pure quartzite, or a mixture of quartzite and limestone, or it may be diabase, or any combination of the three. The iron and copper contents of the ores vary, the iron ranging from 6 to 20 per cent and the copper from 1.5 to 4.0 per cent.

Up to the present no effort has been made to mix these different ores in the proper proportions to obtain the best metallurgical results. It would be impractical to do this in the mine under present conditions. To mix the ore on the surface would require a bedding system such as is used in smelting plants, or a series of bins, one for the ore from each level.

The ore, as a whole, is medium hard to crush and grind; some of it, however, is soft and some is very hard; it tends to become harder as more quartzite is encountered. The average moisture content of ore delivered to the concentrator is 2.5 per cent.

The predominating copper mineral is chalcopyrite, with some bornite and chalcocite and minor quantities of other minerals. A mineralogical analysis of the concentrator heads and a chemical analyses of the same product are given in the following tabulations.

Mineralogical analysis of concentrator heads

Mineral	Per cent of mineral	Per cent of total copper in mineral
Chalcocite	0.15	3.90
Chalcopyrite	6.46	72.69
Bornite	.87	17.95
Covellite	.01	0.23
Cuprite	.18	5.23
Native copper	Trace	- -
Pyrite	10.81	- -
Gangue	81.52	- -
Total	100.00	100.00

Chemical analyses of concentrator heads

	Per cent								Ounces per ton	
	Copper									
	Sulphide	Oxide	Total	Iron	Alumina	Sulphur	Silica	Moisture	Silver	Gold
Year, 1929	2.50	0.11	2.61	14.3	9.0	11.7	49.3	2.5	0.35	0.012
January to May, 1930	2.26	.11	2.37	12.6	9.5	7.9	53.1	2.5	.21	.010

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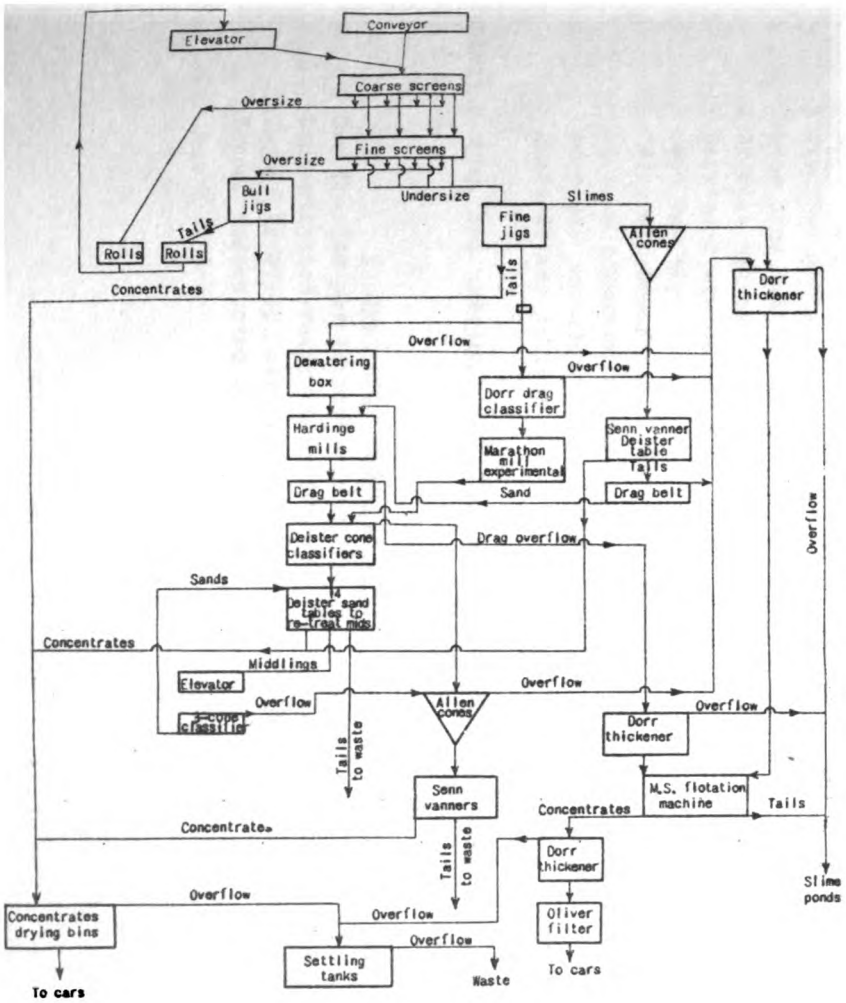

Figure 1.- Concentrator flow sheet, September, 1916

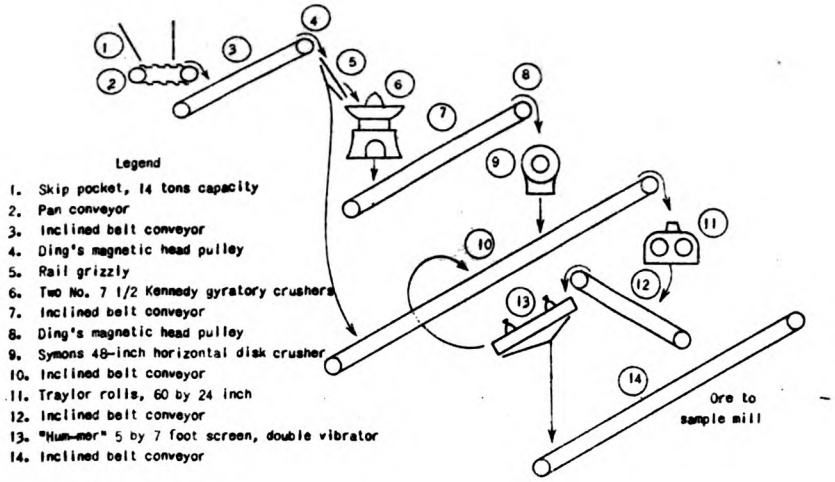
Legend
1. Skip pocket, 14 tons capacity
2. Pan conveyor
3. Inclined belt conveyor
4. Ding's magnetic head pulley
5. Rail grizzly
6. Two No. 7 1/2 Kennedy gyratory crushers
7. Inclined belt conveyor
8. Ding's magnetic head pulley
9. Symons 48-inch horizontal disk crusher
10. Inclined belt conveyor
11. Traylor rolls, 60 by 24 inch
12. Inclined belt conveyor
13. "Hum-mer" 5 by 7 foot screen, double vibrator
14. Inclined belt conveyor
Ore to sample mill

Figure 2.- Coarse crushing plant, unit 1

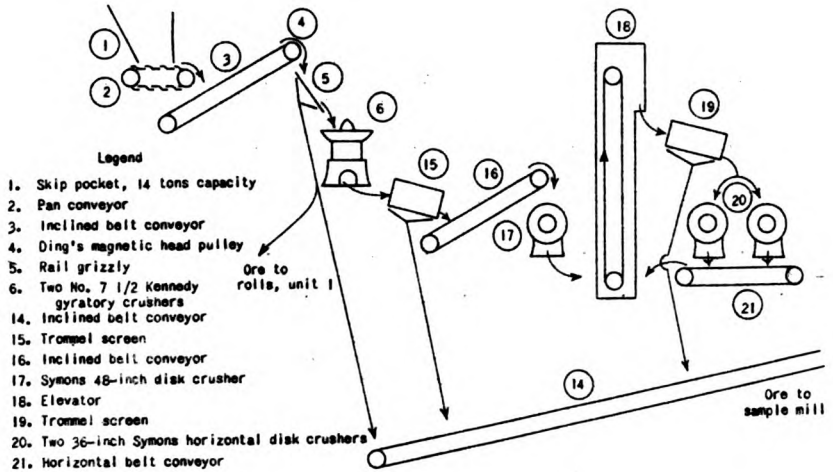
Legend
1. Skip pocket, 14 tons capacity
2. Pan conveyor
3. Inclined belt conveyor
4. Ding's magnetic head pulley
5. Rail grizzly
6. Two No. 7 1/2 Kennedy gyratory crushers
14. Inclined belt conveyor
15. Trommel screen
16. Inclined belt conveyor
17. Symons 48-inch disk crusher
18. Elevator
19. Trommel screen
20. Two 36-inch Symons horizontal disk crushers
21. Horizontal belt conveyor
Ore to rolls, unit 1
Ore to sample mill

Figure 3.- Coarse crushing plant, unit 2

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Physical changes occur in the ore after it stands in the mine, as in shrinkage stopes. The fresh ore responds very well to the flotation treatment, but when the last part of the ore from a shrinkage stope is drawn a marked increase in tailings loss is observed. A similar increase in tailings loss also occurs when the ore is stored in bins or cars for a week. Under these conditions, the sulphide minerals show a rusty, tarnished appearance, although the soluble copper content by assay will be about the same as in the fresh ore.

The minus 200-mesh material in the concentrator feed amounts to about 10 per cent, and this material contains about 0.30 per cent of acid-soluble copper, as compared to 0.10 per cent of acid-soluble copper in the total ore.

HISTORY

The first concentrator erected to treat Old Dominion ores began operations in August, 1905. The capacity of this first plant was approximately 300 tons per day, and gravity methods of concentrating were used throughout. The equipment consisted of the various concentrating machines in general use at that time and included bull jigs, coarse and fine jigs, and Chilian mills, tables, and vanners.

In 1914 a new concentrator was built with a capacity of 800 tons per day. All of the secondary crushing equipment and grinding mills installed for this plant are in use at present. Figure 1 shows the flow sheet, and the following tabulation gives the equipment of this plant as operated in September, 1916.

2 Traylor rolls, 42 by 16 inch.
3 Woodbury jigs, four compartment.
5 Hardinge pebble mills, 8 foot by 36 inch.
1 Long drag belt classifier.
2 Deister hydraulic classifiers.
17 Deister diagonal tables.
2 Senn vanners.
1 "Minerals Separation" flotation machine, 16 cell.
Various accessory machines including shaking screens, settling tanks, thickeners and two small Oliver filters.

Concentrator data and chemical analyses of concentrator products for the 1916 period follow.

Concentrator data for 1916

Ore treated per 24 hours, tons	791
Concentration ratio, tons into 1	2.8
Copper recovery, per cent	86.63
Copper unaccounted for, loss, per cent	1.52

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Chemical analyses of concentrator products for September 1916

	Per cent					
	Copper	Silica	Alumina	Iron	Lime	Sulphur
Feed	3.89	47.5	12.0	14.7	1.5	12.5
Concentrates	9.48	15.3	5.3	30.7	1.0	31.5
Tailings	0.72	- -	- -	- -	- -	- -

In the original concentrator of 1914, and until 1918, the sand tailings product of the Deister tables was rejected as mill tailings. In 1918 one 8-foot by 36-inch Hardinge ball mill was installed to regrind this sand tailings product for flotation treatment. At the same time eight 10-foot Rork flotation machines and sixteen 8-foot Callow flotation cells were installed to replace the 16-cell "Minerals Separation" machine, which had formerly treated the slime product, and to also treat the reground sand product. The 8-foot Callow cells were changed later to 16-foot, flat-bottomed cells.

In 1923 experiments were made for an all-flotation treatment. Until this time, the ore milled contained a considerable amount of native copper which was recovered by jigs and tables. The all-flotation treatment was a success, the first day's operation handling 500 tons of ore and the second day, the full mill capacity of 800 tons.

The reagents used consisted of lime, reconstructed oil, pine oil, and a mixture of thiocarbanilid in orthotoluidine. These changes resulted in a decided improvement in metallurgical results. A comparison of the results obtained by the combined gravity and flotation treatment with those of the all-flotation treatment is as follows:

Comparison of combined gravity and flotation treatment with all-flotation treatment

	Gravity and flotation treatment, July to November, 1923	All-flotation treatment, February to June, 1924
Copper in feed, per cent	4.37	4.34
Concentrates analyses		
Copper, per cent	11.53	22.23
Insoluble, per cent ..	17.4	9.2
Iron, per cent	31.4	30.1
Copper in tailings, per cent	0.41	0.35
Indicated recovery, per cent	93.96	93.41
Concentration ratio, tons into 1	2.81	5.48
Ore treated per 24 hours, tons..	889	867

In 1925 the Forrester matless-type flotation machine was developed and put into operation, replacing the Rork and Callow machines. This change further improved metallurgical results and lowered costs.

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PRESENT COARSE CRUSHING PLANT

The coarse crushing plant, consisting of two units, is located adjacent to the main shaft of the mine. The flow sheet of unit 1, with accompanying legend, is shown in Figure 2.

The ore is hoisted in skips of 3½-ton capacity and dumped into a skip pocket of 14-ton capacity which has sloping sides and a pan conveyor for a bottom. The maximum-sized ore lump is 16 inches and is governed by grizzlies placed over the ore pockets on the different levels of the mine.

From the skip pocket, the ore is fed by a pan conveyor onto a 36-inch inclined belt conveyor equipped with a Ding's magnetic head pulley and thence over a rail grizzly, the oversize product of which goes to one of two No. 7½ Kennedy gyratory crushers, depending on which unit is being used, and at times to both crushers.

When Unit 1 is in service, the gyratory crusher product, minus 4 inches in size, discharges onto a 36-inch inclined belt conveyor, equipped with a Ding's magnetic head pulley and is fed to a 48-inch Symons horizontal shaft disk crusher. The disk crusher product, minus 1½ inches in size, joins the undersize product of the grizzly ahead of the gyratory crusher, on a 24-inch inclined belt conveyor and is carried to a set of 60 by 24 inch Traylor rolls. The roll product drops onto a 24-inch inclined belt conveyor which feeds a 5 by 7 foot double vibrator "Hum-mer" screen, equipped with "Ton-Cap" screen cloth having ¾-inch holes, and operated in closed circuit with the rolls. The screen undersize product is carried to the sample mill by a 24-inch inclined belt conveyor. This unit has a capacity of 125 tons of ore per hour.

The details of machines contained in unit 1, including the methods of driving machines and the details of motors used, are contained in the following tabulation. The numbers used to designate machines in this tabulation correspond to numbers shown in Figure 2.

Details of machines in Unit 1 as shown in Figure 2

(1 to 5) See legend for Figure 2.

(6) No. 7½ Kennedy gyratory crusher driven by motor and belt. General Electric motor, 50-hp., 60-cycle, 2,200-volt, 870 r.p.m. full load, 900 r.p.m. no load; belt is 6-ply and 16 inches wide.

(7) Conveyor belt, 5-ply and 36 inches wide, driven through a gear and pinion connected to a Palmer-Bee No. 160 mill type speed reducer, ratio 27 to 1, by 10-hp. Westinghouse motor operating at 220 volts and 1,160 r.p.m. under full load.

(9) Symons 48-inch horizontal shaft disk crusher driven through short center belt drive to line shaft and thence to crusher pulleys by General Electric 75-hp. slip-ring motor operating at 60 cycles, 2,200 volts, and 490 r.p.m. under full load.

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(10) onveyor belt, 5-ply and 24 inches wide, direct-connected to Palmer-Bee No. 160 mill type speed reducer, ratio 27 to 1 and driven by a 10-hp. Westinghouse motor operating at 220 volts and 1,160 r.p.m. under full load.

(11) Traylor rolls, 60 by 24 inch, each roll is belt driven by a 100-hp. Westinghouse 60-cycle, 3-phase, 2,200-volt motor operating at 575 r.p.m. under full load. Belts used are 20-inch endless, 6-ply, operated with short center idler pulleys.

(12) Conveyor belt, 24 inches wide and 5-ply, direct-connected to Palmer-Bee No. 160 mill type speed reducer, ratio 27 to 1, and driven by 10-hp., 220-volt, Westinghouse motor operating at 1,160 r.p.m. under full load.

(14) Conveyor belt, 24 inches wide and 6-ply, driven by 50-hp. motor through belt connected gear and pinion.

The flow sheet of unit 2, with accompanying legend, is shown in Figure 3. As previously noted, the treatment of mine ore as far as the crushing in gyratory machines is the same for both units.

In unit 2 the oversize product of the rail grizzly passes to one of the No. 7½ Kennedy gyratory crushers and thence to one 8 by 5 foot trommel screen with 1-inch round holes. The undersize product of the grizzly passes to the conveyor belt going to the rolls of unit 1 and sometimes joins the trommel undersize product on the belt which conveys this product to the sample mill. The oversize product of the trommel drops onto a 36-inch inclined belt and is conveyed to a 48-inch Symons horizontal disk crusher. The product from the disk crusher, 1½ to 2 inches maximum size, goes to an elevator which discharges to a 10 by 5 foot trommel screen with 1-inch round holes. The oversize product of the trommel is fed to two 36-inch Symons horizontal disk crushers. The crushed product drops to a 24-inch horizontal belt conveyor which in turn discharges the product into the elevator feeding the trommel, thus placing the disk crushers in closed circuit with the trommel. The undersize product of the trommel joins the other finished products on the 24-inch belt and is conveyed to the sample mill. The capacity of unit 2 is 125 tons of ore per hour.

The details of machines contained in unit 2, including the methods of driving and the details of motors used, are contained in the following tabulation. The numbers used to designate machines in this tabulation correspond to numbers shown in Figure 3.

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Details of machines in Unit 2 as shown in Figure 3

(1 to 5) See legend for Figure 2.

(6) No. 7½ Kennedy gyratory crusher driven by 75-hp., 60-cycle, 2,200-volt General Electric motor through flexible coupling to line shaft. Motor operates at 514 r.p.m. without load and at 490 r.p.m. under full load. Belt driving crusher is 6-ply and 16 inches wide.

(7) Trommel, 8 by 5 foot with 1-inch round holes, driven by 4-ply belt 6 inches wide from the same line shaft that drives the gyratory crusher.

(8) Conveyor belt, 6-ply and 36 inches wide, driven through gear and pinion by 10-hp. motor connected by a 4-ply belt 6 inches wide.

(9, 10, 11, 12 and 13) Symons 48-inch disk crusher, elevator, 10 by 5 foot trommel with 1-inch round holes, two 36-inch Symons disk crushers and 24-inch belt conveyor all driven by a 150-hp., 60-cycle, 2,200-volt General Electric motor which operates at 720 r.p.m. without load and at 700 r.p.m. under full load. This arrangement is unsatisfactory and steps are being taken to remedy this condition. The new arrangement is as follows: Line shaft A driven at 202 r.p.m. by a 150-hp. motor operating at 700 r.p.m. Motor pulley 22-inch diameter and 21-inch face, driven pulley 76-inch diameter and 20½ inch face, belt 6-ply and 18 inches wide with speed of 4,031 feet per minute. Counter shaft B driven at speed of 202 r.p.m. from line shaft A. Line shaft pulley is 16-inch diameter and 17-inch face and is operated by a clutch. The countershaft pulley is 68-inch diameter and 17-inch face.

(9) Symons 48-inch disk crusher is driven from the counter shaft B. The driving pulley for the inner shaft of the crusher is 68-inch diameter and 16½ inch face and drives the pulley of the inner shaft which is 54-inch diameter and 16½ inch face at speed of 254 r.p.m. through a 6-ply belt 14 inches wide. The outer shaft of the crusher is driven from countershaft B at speed of 90 r.p.m. through a 6-ply belt, 16 inches wide. The driving pulley has 24-inch diameter and 16½ inch face and the driven pulley has 54-inch diameter and 19-inch face.

(10) The elevator is driven from countershaft C, which in turn is operated from countershaft B. The driving pulley on countershaft B has a 32-inch diameter and 12½ inch face and drives the pulley on countershaft C, which has 50-inch diameter and 12½ inch face, at speed of 129 r.p.m., through a 6-ply belt 12 inches wide. The drive from countershaft C to the elevator is through a 4-ply belt 8 inches wide. The driving pulley on countershaft C has 34-inch diameter and 8½ inch face and the driven pulley, 40 inches in diameter and 8½ inch face, operates at speed of 110 r.p.m.

(11) The 10 by 5 foot trommel is driven from countershaft C through a 4-ply belt, 10 inches wide. The driving pulley has 26-inch diameter and 10-inch face and the driven pulley, 60 inches in diameter and 11-inch face, operates at speed of 56 r.p.m.

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(12) The two 36-inch Symons disk crushers are driven from line shaft A. The driving pulley for the inner shaft of the crusher has 66-inch diameter and 14½-inch face and the driven pulley, 44-inch diameter and 14½-inch face, operates at speed of 303 r.p.m. through a 6-ply belt, 12 inches wide. The driving pulley for the outer shaft of the crusher has 29-inch diameter and 14½ inch face and the driven pulley, 44-inch diameter and 14½-inch face, operates at speed of 133 r.p.m. through a 6-ply belt, 14 inches wide.

(13) Belt conveyor is driven from line shaft A through a 4-ply belt, 6 inches wide. The driving and driven pulleys both have 24-inch diameters and 6½-inch faces, and a speed of 202 r.p.m.

The product of the coarse crushing plant is weighed enroute to the sample mill by a Merrick weightometer.

Screen analyses of the products of coarse crushing units 1 and 2, are given in Table 1.

SAMPLE MILL

The sample mill was built at the same time as the coarse crushing plant and concentrator, in 1914. At that time the Old Dominion Co. treated custom ores from the mines and leases of the district both at the concentrator and at the smelter. At present all ore and lime passing through the crushing plant is sampled in the sample mill.

Figure 4 shows the flow sheet of the sample mill and is accompanied by a legend giving the details of the equipment used.

The ore is conveyed from the coarse crushing plant to the sample mill and drops into the mouth of a chute that leads to the fine ore bins. The No. 1 Vezin sample cutter passes through the stream of ore at this point at the rate of 13 times per minute and cuts out a sample amounting to approximately 10 per cent of the total ore.

The sample passes through a mixing drum revolving at 12 r.p.m. and is then fed to No. 2 sample cutter which makes a 20 per cent cut. The sample from the No. 2 cutter is passed by chute to a set of 36 by 16 inch rolls and the crushed product fed to No. 3 cutter, which makes a 20 per cent cut. The sample from No. 3 cutter is again crushed by a set of 28 by 14 inch rolls, the crushed product going to No. 4 cutter which cuts out a 20 per cent sample. This sample is again crushed in a set of 12 by 12 inch rolls and the crushed product passed to two No. 5 cutters, each cutter making a 10 per cent cut. The No. 5 cutters produce original and duplicate samples, each of which goes to a separate hopper. The final samples are further reduced and prepared in the sample room.

The rejects of Vezin cutters 1 and 2 are conveyed by chute and the rejects of cutters 3, 4, and 5 by elevator to the concentrator storage bins.

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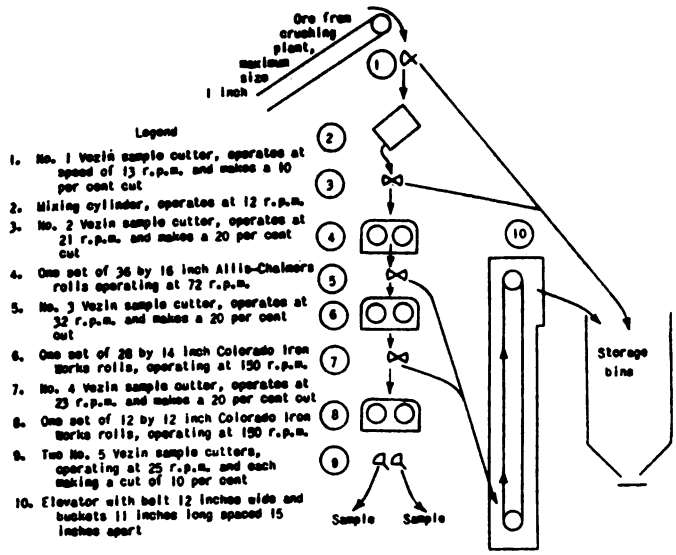
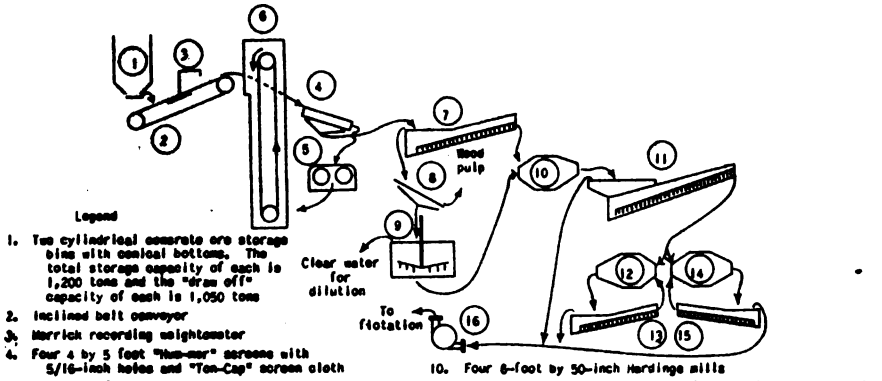


Legend

1. No. 1 Vezin sample cutter, operates at speed of 13 r.p.m. and makes a 10 per cent cut
2. Mixing cylinder, operates at 12 r.p.m.
3. No. 2 Vezin sample cutter, operates at 21 r.p.m. and makes a 20 per cent cut
4. One set of 36 by 16 inch Allis-Chalmers rolls operating at 72 r.p.m.
5. No. 3 Vezin sample cutter, operates at 32 r.p.m. and makes a 20 per cent cut
6. One set of 28 by 14 inch Colorado Iron Works rolls, operating at 150 r.p.m.
7. No. 4 Vezin sample cutter, operates at 23 r.p.m. and makes a 20 per cent cut
8. One set of 12 by 12 inch Colorado Iron Works rolls, operating at 150 r.p.m.
9. Two No. 5 Vezin sample cutters, operating at 25 r.p.m. and each making a cut of 10 per cent
10. Elevator with belt 12 inches wide and buckets 11 inches long spaced 15 inches apart

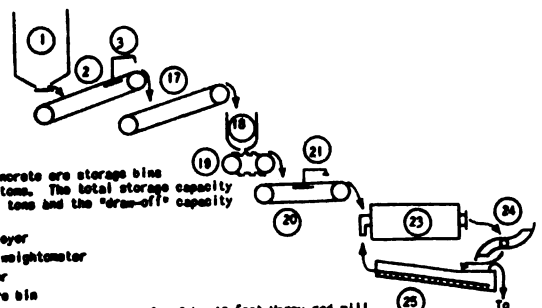
Figure 4.- Flow sheet of sample mill



Legend

1. Two cylindrical concrete ore storage bins with conical bottoms. The total storage capacity of each is 1,200 tons and the "draw off" capacity of each is 1,050 tons
2. Inclined belt conveyor
3. Merrick recording weightometer
4. Four 4 by 5 foot "Hum-mer" screens with 5/16-inch holes and "Ton-Cap" screen cloth
5. Two sets of 43 by 16 inch Traylor rolls
6. Two bucket elevators, belts 20 inches wide
7. Three 4 1/2 by 17 foot Dorr duplex classifiers
8. One chip screen
9. One 40-foot Dorr thickener
10. Four 8-foot by 50-inch Hardinge mills
11. One 8 by 23 foot Dorr bowl classifier, 18-foot diameter bowl
12. One 6-foot by 50-inch Hardinge mill
13. One 6 by 20 foot Dorr duplex classifier
14. One 8-foot by 36-inch Hardinge mill
15. One 8 by 20 foot Dorr duplex classifier
16. One 6-inch Wilfley sand pump

Figure 5.- Flow sheet of old unit for fine crushing and grinding



Legend

1. Two cylindrical concrete ore storage bins with conical bottoms. The total storage capacity of each is 1,200 tons and the "draw-off" capacity is 1,050 tons
2. Inclined belt conveyor
3. Merrick recording weightometer
17. Cross belt conveyor
18. Suspended steel ore bin
19. Two pan feeders
20. Horizontal belt conveyor
21. Merrick weightometer
23. One 6 by 12 foot Marcy rod mill
24. Scoop elevator
25. One 8-foot by 26-foot 8-inch Dorr duplex classifier

Note:-Items 1, 2, and 3 serve both grinding units of the concentrator

Figure 6.- Flow sheet of new or Marcy mill unit

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The original sample is thoroughly mixed and the moisture determination is made on this sample as soon as the run is completed. The residue from the moisture determination is returned to the original sample and the entire sample is reserved for making up a monthly composite sample. The duplicate sample is used for the assay of each lot, or the ore from each level, and also for the daily composite sample of the different mine levels.

The weight of the sample has been checked against the tons of ore sampled so many times that it is possible, when the weightometer is out of order, to estimate very closely the number of tons of ore sampled by the weight of the sample. The sample of ore from each level in the mine is kept separate.

PRESENT METHOD OF CONCENTRATING

The present concentrator consists of two grinding units known as "the old unit" and "the new unit" and one flotation unit. The capacity of the old unit is approximately 1,200 to 1,250 tons per 24 hours and that of the new unit is 275 tons per 24 hours. The new unit was installed to provide additional capacity and also to serve as a test unit in determining the best flow sheet and arrangement of equipment for remodeling the entire concentrator.

The new unit grinds the product of the coarse crushing plant for flotation treatment in a Marcy rod mill operating in closed circuit with a Dorr classifier. The old unit prepares the product of the coarse crushing plant for flotation treatment by crushing in rolls and then by two stages of grinding in Hardinge ball mills; the second stage operates in closed circuit with Dorr classifiers.

THE OLD CRUSHING AND GRINDING UNIT

Figure 5 presents a flow sheet of fine crushing and grinding in the older unit and is accompanied by a legend giving the details of equipment used.

Intermediate Crushing

The ore is fed from the concentrator storage bins by means of Challenge feeders onto an inclined 18-inch belt conveyor operating at a slope of 13° and at a speed of 127 feet per minute. A deflector located at the head end of this belt and operated by a cam shunts a portion of the feed amounting to 275 tons per day onto a cross conveyor which discharges this material to a storage bin serving the new Marcy mill grinding unit. The remaining feed, amounting to 1,225 tons per day, is conveyed by launder to four 4 by 5 foot "Hum-mer" screens equipped with "Ton-Cap" screen cloth having 5/16-inch openings. The oversize product feeds two sets of 43 by 16 inch rolls, the crushed roll products going to two 20-inch bucket elevators and thence to the "Hum-mer" screens. The rolls and elevators therefore operate in closed circuit with the "Hum-mer" screens the circulating load amounting to approximately 100 per cent. This intermediate crushing circuit operates wet, the water being introduced under the rolls and at the head of the main feed launder following the deflector. The rolls are belt-driven from a line shaft operated by one synchronous motor. This motor also operates the elevators and the main feed conveyor, which are driven by belts from countershafts.

The undersize product of the "Hum-mer" screens is the final product of intermediate crushing and furnishes the feed to the primary grinding unit. A screen analysis of this product is given in Table 2.

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Primary Grinding

The undersize product of the Hum-mer screens contains an excess of water which is removed by three 4½ by 17 foot, model-C, Dorr classifiers. The overflows from these classifiers, amounting to about 10 per cent of the feed to the primary grinding section, contain the primary slime and also the wood chips in the feed pulp. These overflow products are fed to a chip screen which removes the wood pulp and thence to a 40-foot Dorr thickener. The thickener overflow product is used as diluting water and the thickened pulp joins the sand products of the dewatering classifiers as feed to the primary Hardinge mills. These sand products of the dewatering classifiers contain 70 per cent of solids and are conveyed from the classifiers to a distributing box feeding the four Hardinge mills by launders which have a slope of 9 inches per foot.

The four primary Hardinge mills were originally 8-foot by 36-inch pebble mills but were converted to 6-foot by 50-inch Hardinge ball mills by lagging with 10-inch timbers and lining with cast-iron liners. Each mill is operated by a 75-hp. motor which consumes about 80-hp. input when operating at a speed of 490 r.p.m. Each mill operates with a charge of 13,000 pounds of 2½-inch cast-iron balls. The balls and liners are cast at a local foundry. The ball consumption for the year 1929 amounted to 1.32 pounds per ton of ore and the liner consumption was 0.22 pound per ton for the same period. These mills may eventually be replaced by 6½ by 12 foot Marcy rod mills.

The primary ball mill product contains 50 per cent of solids and is the feed to the bowl classifier of the regrinding unit. A screen analysis of this product is given in Table 2.

Regrinding

The equipment for regrinding consists of one 8 by 23 foot Dorr bowl classifier equipped with an 18-foot diameter bowl, one Hardinge mill which was converted from an 8-foot by 36-inch pebble mill to a 6-foot by 50-inch ball mill by lagging with timbers as described for the primary mills and one 8-foot by 36-inch Hardinge ball mill.

The ground products of the primary Hardinge mills are conveyed by launders, which have a slope of 2 inches per foot, to the bowl classifier, the bowl rakes of which operate at a speed of 4 r.p.m. and the drag rakes at 30 strokes per minute.

The sand product of the Dorr bowl classifier containing 79 per cent of solids is split by deflectors in front of the classifier, and each portion is conveyed to its respective secondary Hardinge mill by launder.

The 6-foot by 50-inch Hardinge mill is driven by a 75-hp. motor connected to the pinion shaft of the mill by a flange flexible coupling. This mill is in closed circuit with a 6 by 20 foot Dorr duplex classifier operating at a slope of 3¼ inches per foot and with a rake speed of 28 strokes per minute.

The 8-foot by 36-inch mill is driven by a 150-hp. motor operating at a speed of 430 r.p.m. under full load and connected to the pinion shaft of the mill through a Francke flexible coupling. This mill is in closed circuit with an 8 by 20 foot Dorr duplex classifier operating at a slope of 3¼ inches per foot and with a rake speed of 28 strokes per minute.

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The reason for the high speed of the classifier rakes is due to the excessive amount of pyrite in the secondary grinding circuit, this mineral concentrating at this point in the flow sheet.

Cast-iron liners and 2-inch cast-iron balls are used in the secondary grinding mills; both liners and balls are cast at a local foundry. The liner consumption amounted to 0.08 pound and the ball consumption to 1.28 pounds per ton of ore ground for the year 1929.

The ground pulp discharged from the 6-foot by 50-inch mill contains about 60 per cent of solids and that from the 8-foot 36-inch mill about 70 per cent of solids. The circulating loads in the secondary grinding circuits amount to approximately 360 per cent. The feed to the regrinding or secondary ball mills is estimated to be 50 per cent of the total ore fed to the "old" grinding unit.

The sand products of the Dorr classifiers in circuit with the ball mills are returned to their respective mills by launders having slopes of 6 inches per foot.

The overflow product of the bowl classifier joins the overflow products of the drag classifiers, which contain 27 per cent of solids, and the combined products are conveyed by launder to a 6-inch Wilfley sand pump which in turn delivers this material to flotation treatment.

Table 3 gives screen analyses of the feed and products of the bowl classifier and also screen analyses of the feed and products of the classifiers in closed circuit with the secondary or regrinding mills.

The total power consumed by the old unit in reducing 1 ton of old unit feed to flotation size amounts to 10.63 kw.h. The power consumed per ton of old unit feed is distributed as follows:

	Kilowatt hours
Rolls, elevators, and screens	1.35
Primary grinding	4.97
Secondary grinding, treats 50 per cent of total feed	4.31
Total	10.63

THE NEW GRINDING UNIT

The new or Marcy mill unit was installed in 1926. As previously stated, this unit was added to obtain additional capacity and also to serve as a test section. As a test section it was used primarily in determining a flow sheet for future remodeling and incidently for testing the numerous flotation reagents that were being developed at that time.

Figure 6 gives a flow sheet of this grinding unit and the legend accompanying the flow sheet gives the details of equipment used.

The ore from the main concentrator storage bins is conveyed to the top of the mill building and enroute passes over a Merrick weightometer, the conveyor and weightometer serving both grinding units of the concentrator. At the discharge of this conveyor, the feed

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for the new grinding unit is cut periodically from the main ore stream by a deflector operated by a cam attached to the head pulley of the conveyor. The ore cut from the main stream, amounting to 275 tons per 24 hours, drops to a belt conveyor and is discharged into a surge bin serving the Marcy unit.

From the surge bin the ore is fed onto a conveyor belt by a pan feeder, the conveyor belt feeding one 6 by 12 foot Marcy rod mill. The ore is weighed enroute to the Marcy mill by a Merrick weightometer.

The rod mill is driven at a speed of 17½ r.p.m. by a 150-hp. motor operating through a Poole Turbo gear reduction unit, the motor consuming about 130-hp. under full load. The rod load amounts to between 40,000 and 42,000 pounds of 3-inch by 12-foot steel rods, each rod weighing approximately 300 pounds. The rod mill ground product is discharged into a scoop elevator which raises it to an 8-foot by 26-foot 8-inch Dorr duplex classifier, operating in closed circuit with the rod mill.

The scoop feeder is chain driven at a speed of 17 r.p.m. by a 7½-hp. motor operating through an IXL worm-gear speed reducer, type 27-W, with reduction ratio 36 to 1. This reducer was installed to replace a type-15W IXL reducer operated by a 3-hp. motor as the smaller reducer could not satisfactorily handle the unusual loads which would come at times.

The classifier operates at a slope of 3 inches per foot, a rake speed of 31 strokes per minute, and with a circulating load amounting to approximately 300 per cent. It is operated through a "Tex-rope" drive by a 7½-hp. motor. The classifier overflow product, containing 20 per cent of solids, goes directly to the rougher flotation cells. This product, however, will go to the bowl classifier when the additional secondary grinding mills now contemplated are installed. The new grinding mills will probably be installed before the end of 1930.

The total power consumed per ton of ore treated in the new unit amounts to 7.39 kw.h., as compared to 10.63 kw.h. per ton of ore handled by the old unit. Table 4 gives screen analyses of the feed and products of the new grinding unit.

FLOTATION

Figure 7 gives a flow sheet of flotation treatment and is accompanied by a legend giving details of equipment used.

The flotation feed, amounting to 1,500 tons of solids per 24 hours, consists of the overflow products of the bowl classifier, the classifier in circuit with the secondary or regrinding mills of the old unit and the classifier in circuit with the Marcy mill of the new unit. The overflow products of the bowl and regrinding mill classifiers, containing approximately 22 per cent of solids, are elevated 52 feet and discharged to a distributing box at the head of the two 10-foot primary Forrester flotation cells by a 6-inch Wilfley sand pump. The overflow product of the classifier in circuit with the Marcy mill flows into the same box by gravity.

The two primary flotation cells operate in parallel and each treat 750 tons of solids per 24 hours. The cells recover approximately 65 per cent of the total copper in the feed and produce finished high-grade concentrates, which average between 25 and 30 per cent

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of copper, and tailings which are fed to the secondary flotation cells. The power consumed by the primary cells amounted to 0.0814 kw.h. per ton of ore treated during 1929.

The two 17-foot secondary flotation cells, which treat the tailings products of the primary cells, operate in parallel and produce finished concentrates containing approximately 20 per cent of copper. The power consumption of the secondary flotation cells amounted to 0.1386 kw.h. per ton of ore treated during 1929. The tailings products of the secondary cells are distributed to four 15-foot tertiary cells operating in parallel.

The tertiary flotation cells produce concentrates containing from 10 to 12 per cent of copper. Attempts to increase the grade of the tertiary flotation concentrates by separate treatment in a cleaner cell prior to 1929 were not successful and, during the year 1929, these products were added to the finished concentrates until provision could be made for additional treatment. At present these concentrates are pumped to the secondary Hardinge ball mills where they are subjected to further grinding and conditioning. The regrinding of tertiary flotation concentrates has resulted in increasing the average grade of total concentrates produced from 18.97 per cent of copper in 1929 to 24.43 per cent of copper for the first five months of 1930. The tertiary flotation cells consumed 0.60 kw.h. per ton of ore treated in 1929. The tailings products of the tertiary cells are treated in four 17-foot scavenger cells.

The scavenger cells operate in parallel and produce final tailings products and low-grade concentrates. These concentrates, containing from 1 to 2 per cent of copper, are treated in two 10-foot intermediate cleaner cells producing concentrates which join the tertiary concentrates for regrinding, and tailings which are pumped to the head of the tertiary cells. The power consumed by the scavenger cells amounted to 0.73 kw.h. per ton of ore treated in 1929.

The final flotation tailings are sampled by automatic samplers and are then pumped through 4,000 feet of 8-inch pipe line to the tailings pond.

The kinds and amounts of flotation reagents used are as follows:

	Pounds per ton of ore
Lime, added as milk of lime	9.005
Potassium ethyl xanthate	.081
Reconstructed oil	.088
Pine oil	.193

The reconstructed oil is a mixture containing 87.5 per cent, by weight, of Scotch blast furnace oil and 12.5 per cent of pine oil, to which sulphur is added in quantity amounting to 6 per cent of the weight of the oil mixture. The mixture is boiled in a reflux still at temperature of 215 to 225° F. for about eight hours.

The lime is obtained from a quarry located on Old Dominion property. The quarry is operated by a leasee who burns the lime for local consumption. The burned lime is received in 40-ton capacity steel hopper-bottom cars and dumped into a bin at the head of the crushing plant. It is reduced to ¾-inch size in the crushing plant and delivered by belt conveyors to a suspended steel storage bin of 150 tons capacity. From the storage bin the lime is fed by gravity to a 4 by 8 foot Marathon mill, operated by a 50-hp. motor connected

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to the pinion shaft of the mill by a Morse chain drive. The mill uses 2-inch balls and water is added to the lime in the mill feed box. The milk of lime product from the mill is conveyed by launder to a 28-foot diameter storage tank, 10 feet high, equipped with a paddle agitator operated at a speed of 10 r.p.m. by a 10-hp. motor, for keeping the solids in suspension. From this tank the lime is circulated in the concentrator through a 2-inch pipe line by a 4-inch Krogh pump, belt-driven by a 15-hp. motor, the excess product discharging into the top of the storage tank through a screen. Draw-off valves are placed in the circulating line at convenient points and the quantity of milk of lime fed to the various pulps is regulated by varying the size of nipples in the draw-off valves. The milk of lime is fed to each grinding mill and to the thickener of the old grinding unit.

Flotation oils are added to the regrinding Hardinge mills and to the Marcy mill, and at times pine oil is added to the secondary and tertiary flotation cells. Xanthate is added to the pulp of the old grinding unit at the flotation feed pump and to the Marcy classifier overflow product of the new unit. Additional xanthate is added to the pulps of the secondary and tertiary flotation cells.

The air used for flotation is obtained from one No. 6 and three No. 5 Roots blowers, the same blower equipment formerly serving the Callow cells which were replaced by Forrester cells. One of the No. 5 blowers furnishes air for the two primary and the two secondary cells and the other blowers furnish air for the four tertiary, the four scavenger, and the two intermediate cleaner cells. The No. 6 blower and two of the No. 5 blowers are driven from a line shaft by one 175-hp. motor, this motor also serving the feed, concentrates, and middlings sand pumps. This arrangement is unsatisfactory from the standpoint of both operation and of safety, and steps are being taken to change it. The total volume of air supplied to flotation operations amounts to 9,500 cubic feet of free air per minute, the pressure being 2 and 1½ pounds per square inch at the blower and flotation cells respectively.

The Forrester flotation machines were adopted, after extensive and thorough testing, because of ease of operating, lower operating and repair costs, and improved metallurgical results as compared to competing machines tested.

After the adoption of the all-flotation treatment of ores in 1924 with the use of lime and T-T mixture reagents it became evident that a change would have to be made in the machines using porous blankets. The blankets of these machines became clogged with lime salts in a short time and extra men were required to pound the mats in order to keep them open. The rejected pyrite settled on the blankets, which further impeded aeration of the pulp. The air pressure was increased from 4 to 6 and 7 pounds per square inch and even with this increase of pressure it was difficult to get sufficient air through the blankets for the proper aeration of the flotation pulps. It became necessary to change one blanket each day, and as there were 16 cells in operation the life of a blanket was 16 days. Furthermore, in order to obtain this 16-day life it was necessary to give each blanket an acid bath and a scrubbing every four days. Operating conditions at that time for pneumatic machines using porous blankets were, therefore, not satisfactory at the Old Dominion concentrator and it was necessary to either adopt mechanical agitation machines or to devise some means of producing a froth other than by passing air through a porous medium. It was due to these difficulties that the Forrester method and machine were developed. The Forrester machine, as is generally known, is a matless-type pneumatic machine, the froth being produced by air, at low pressure, introduced through ¼-inch vertical parallel pipes spaced on 4-inch centers.

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After the development of this new method and machine, tests were conducted to compare the Forrester-type machine with a revolving porous-medium type of machine and it was tered with the stationary blanket type of pneumatic cell.

Comparative tests were then run between the Forrester-type machine, the Rork and double-spitz K and K mechanically agitated machines, the latter machine having been operated in the Old Dominion concentrator in 1923.

The results of these comparative tests showed that for the treatment of Old Dominion ores the Forrester machine was superior to the mechanically agitated machines tested, the Forrester machine producing better metallurgical results and being more economical in power requirements.

Until this time, the best metallurgical results in the treatment of Old Dominion ores had been obtained by using first a mechanically agitated machine and then a pneumatic machine with porous blankets. The Forrester machine, operating alone on Old Dominion ore, was now tested against two mechanical-pneumatic type combinations. The first combination tested was the K and K cell followed by the Callow cell, and the second was the K and K cell followed by the Forrester cell. The best metallurgical results and the lowest power and maintenance costs were obtained with all-Forrester machine treatment.

Upon completion of the comparative tests described it was decided to adopt the Forrester method and machine at the Old Dominion concentrator. The results of the test work noted at the Old Dominion plant have been substantiated by tests at other well-known concentrators.

A comparison of data obtained by flotation treatment with Forrester machines and by treatment with the Rork-Callow machines is given in Table 5.

Primary slimes are not given a separate flotation treatment at present, although laboratory investigation of separate sand and slime treatments indicate some improvement in results by the separate treatments.

Table 6 gives average analyses of concentrator feed and final products for the year 1929 and for the first five months of 1930. Table 7 shows the distributions of copper, by screen sizes, in the concentrator feed and final products for May 1930. Table 8 gives mineralogical analyses of concentrator feed and final products for December, 1928, obtained by the examination of polished surfaces with the microscope. This table also shows the per cent recoveries of the separate copper minerals.

DEWATERING AND HANDLING OF CONCENTRATES

Figure 8 is a flow sheet of the system used for the dewatering and handling of concentrates and is accompanied by a legend showing the details of equipment used.

All finished flotation concentrates products are conveyed to the dewatering unit through a 4-inch pipe line 100 feet long, by a 4-inch Wilfley sand pump operating with a 30-foot lift. The flotation concentrates pulps are fed to a 36-inch Akins classifier operated at a speed of 3 r.p.m. and at a slope of 2 inches per foot. The classifier sand product, which contains the rapid settling heavier portion of the total concentrates, discharges into one of nine cylindrical concrete storage bins, 24 feet in diameter by 8 feet

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high. The classifier is mounted on a movable platform which runs over a track extending the full length of the storage bins.

These bins were formerly used for the storage of jig and table concentrates and each has a capacity of approximately 100 tons of dry concentrates. The bins are equipped with filter bottoms constructed with slats covered with cocoa matting, which in turn is covered with canvas. The bottom of each bin is connected by a 2-inch pipe line to a No. 2 Nash Hytor vacuum pump which maintains a constant vacuum amounting to 7½ inches of mercury under the filter bottom of the bins when being drained. The sand product of the Akins classifier, containing about 30 per cent of moisture, remains in the bins for a period of 4 to 6 days, and during this period drains to a product containing 7.5 per cent of moisture. The bins are unloaded directly into railroad cars by a Blaisdell excavator which is mounted to run over the track that serves the classifier platform. These concentrates are known at the plant as "bin concentrates" and comprise 45 per cent of the total concentrates shipped.

The overflow product of the Akins classifier and the overflow of the concentrates bins, which average 12 per cent of solids, are conveyed to one 40-foot Dorr tray thickener through a 4-inch pipe line, 250 feet long, by a 4-inch Wilfley sand pump operating with a 50-foot lift. The overflow product of this thickener passes to an ordinary 40-foot Dorr thickener, the overflow of the second thickener passing to two large settling ponds, which are emptied once each year.

The thickened pulps of the Dorr tanks, which average about 60 per cent of solids, are filtered by a 14 by 14 foot Oliver filter which operates two 8-hour shifts each day. The filter cake, which averaged 16.23 per cent of moisture during the year 1929, drops directly into railroad cars. The filter is washed and cleaned each day and is given an acid bath periodically. It is served by a 16 by 12 inch Alberger dry vacuum pump which maintains a vacuum averaging 22½ inches of mercury. The air for the "blow" is supplied by compressors installed at the central power plant, the blowing air having a pressure of 10 pounds per square inch at the filter. The filter cloth used is 17.5-ounce Palma twill, made into 30-inch widths, lapped ¾-inch, and double-stitched. The average life of filter covers is three months, and each cover handles approximately 4,500 tons of concentrates.

The actual filtering time averages 10 hours per day and the dry weight of the concentrates filtered per day for the year 1929 averaged 91 tons.

The total power consumed in dewatering and handling concentrates during 1929 amounted to 6.33 kw.h. per ton of dry concentrates handled, which is equivalent to 0.78 kw.h. per ton of ore concentrated. These figures include the power consumed by the two 4-inch Wilfley sand pumps, the two vacuum pumps, the Akins classifier, the thickeners, and the filter.

DISPOSAL OF TAILINGS

The disposal of tailings from the Old Dominion concentrator is a rather important problem for the reason that suitable space for this purpose is scarce. The tailings ponds to date have been built on the side of a hill with a long dam and a small area behind the dam.

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The weight of tailings to be handled, amounting to approximately 40,000 tons per month, is not very large as compared to the other concentrators of the Globe-Miami district, but this amount requires 16 acre feet of space per month. As it is necessary to build 2,400 feet of dam in order to obtain a pond area of 15 to 20 acres, the disposal of tailings at low cost presents a problem.

Figure 9 is a flow sheet of tailings disposal. The flotation tailings pulp, containing approximately 20 per cent of solids, flows to a 6-inch Wilfley sand pump which delivers it to two Dorr duplex classifiers through an 8-inch pipe line, 4,000 feet long and approsimately level at the present time. The pump is driven by belt at a speed of 1,143 r.p.m. by a 75-hp. motor and handles an average of 1,333 tons of dry tailings per 24 hours with a power consumption of 0.82 kw.h. per ton of tailings handled. The bowls or cases of the pump last 80 days, the runners 26 days, and the follower plates 60 days.

The two 4½ by 23 foot, model C, Dorr duplex classifiers, which receive the tailings product, are located on the dam and are mounted on a carriage that can be moved over rail sections either forward or backward. They operate at a slope of 2 inches per foot and the rakes are driven at a speed of 10 strokes per minute by a 7½-hp. motor.

The sand products of the classifiers are used to build the dam, and the overflow products are allowed to settle in the pond behind the dam. Clear water from the pond is discharged through spillways and flows down the creek by gravity.

The building of tailings dams by the use of drag classifier sand products was first tried, as an experiment, in 1922. The system proved so much more satisfactory and economical, as compared to the former system of launders and cones, that it has been continued since that time.

In building the dam, the sand products of the classifiers are discharged 20 to 25 feet ahead of the classifiers, forming a bank of material which, after leveling, is 16 feet wide at the top. A mat, made of 2 by 12 inch planks, is laid on the leveled top. These operations are continued until the dam has been extended one length of pipe or a distance of 20 to 22 feet. The tailings pulp feeding the classifier is then diverted temporarily to the pond. In the event that it is impossible to move the classifiers at this time it is possible to carry the sand product further ahead of the normal distance by raising the overflow lip of the classifier, thus diluting the sand product and permitting it to run further. Suitable lengths of rail are spiked to 4 by 6 inch timbers which are laid on top of the 2 by 12 planks ahead of the classifiers. The classifier feed pipes are disconnected and the carriage carrying the classifiers is moved far enough ahead, by pinch bars, to permit another length of pipe to be added to the feed line. A 10-foot length of 8-inch suction pipe is used to connect the feed pipe of the classifiers with the pump line. After the classifiers have been moved and connected to the pump line they are leveled and ready to receive feed. The operation of moving the classifiers takes an hour to an hour and a half.

The dam is built level until within 500 feet of the end and then finished with an upward slope of 6 inches in a pipe length or 20 feet. This is done in order to give the classifiers enough elevation for starting the return trip and thus avoids the necessity of jacking up the classifiers for a vertical distance of approximately 10 feet. At the end of the dam the width is increased to provide sufficient area to turn the classifier carriage

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for the return trip. The lifts average 10 feet except when going over gullies, where they have been built to maximum heights of 50 feet.

The cost of tailings disposal, by the system described, amounts to $0.025 per ton of ore concentrated, including the cost of pumping the tailings pulp to the dam. The former method employed, comprising launders and cones, cost $0.04 to $0.06 per ton of ore concentrated and these figures did not include any charge for pumping because when this system was used the tailings pulp flowed to the impounding ponds by gravity.

Table 9 gives the screen analyses of feed and products of the classifiers operating at the tailings dam obtained from a three-day test.

SAMPLING

A sample of the concentrator feed is taken as explained in the description of the sample mill. Another daily sample of the concentrator feed is taken by hand from the ore stream as it discharges from the main ore conveyor at the head of the concentrator. The monthly results from this latter sample check closely the results obtained from the monthly composite obtained in the sample mill. The results from the sample mill composite, however, are used for all metallurgical calculations.

The concentrates products are sampled by hand for shift samples. These products are again sampled as they are loaded into cars for shipment and are sampled for the third time when the cars are unloaded at the smelter. The results from the three samples check very closely.

Tailings samples are cut in duplicate by two automatic samplers operated by air and controlled by a timing motor and solenoids. The tailings samplers are placed in series, one behind the other, in order to obtain check samples. When the assay results from these samples differ, the average of the two is taken as the final assay.

The unaccounted-for loss in copper for the year 1929 amounted to 0.73 per cent.

The following method of determining acid-soluble copper content of heads and tailings products is used at the Old Dominion concentrator.

A sample weighing 2 grams is covered with water in a beaker and the metallic iron is removed with an electromagnet. The sample is then treated with 50 c.c. of dilute sulphuric acid solution, 5 per cent by volume, saturated with sulphur dioxide and allowed to stand at room temperature for one hour.

The sample is then filtered and the filtrate evaporated to dryness. The residue is dissolved by boiling in water to which 3 c.c. of nitric acid has been added, diluted with cold water, and the copper content determined by electrolysis.

WATER SUPPLY

As previously stated, all of the water used in the concentrator is pumped from the mine and at times is very muddy because of the precipitation of iron salts in the storage tank.

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The temperature of the water is about 110° F. A chemical analysis follows:

	Grams per liter
Insoluble	0.034
Fe_2O_3 and Al_2O_3	.074
CaO, total	.138
CaO, permanent	.112
MgO	.076
S	.126
Suspended matter	.040
Total solids	.900

The total water used per ton of ore ground in 1929 amounted to 1,105 gallons. This was all new water as no water is reclaimed from the tailings pulp.

The question has often been asked by visitors whether or not the hot water has any effect upon the metallurgical results. It is impossible to answer this question, as it is not possible to make a comparison on an operating scale.

METALLURGICAL DATA AND COSTS

Table 10 gives concentrator metallurgical data for the year 1929 and for the first five months of the year 1930. Table 11 gives a summary of direct concentrating costs for the year 1929. Table 12 shows concentrator costs in units of labor, power and supplies for the year 1929.

CONVEYING DATA

Table 13 gives the operating details of belt conveyors and elevators. In the following description of conveyors and in Table 13 the numbers used to designate machines correspond to the numbers given in the various flow sheets.

CRUSHING PLANT CONVEYORS

Pan conveyor 2, shown in Figures 2 and 3, serves as the bottom of the skip pocket at the main hoisting shaft. It is driven from the tail pulley of conveyor 3 and is equipped with a clutch operated by air from the top landing at the shaft. The operating details of this conveyor are given in Table 13.

Belt conveyor 3, Figures 2 and 3, handles run-of-mine ore from pan conveyor 2 to the gyratory crusher. It is driven by a 25-hp. General Electric, variable-speed, shunt-wound, direct-current motor connected by belt to a Dings magnetic pulley which is the head pulley of the conveyor. See Table 13 for operating details.

Belt conveyor 7, Figure 2, conveys the discharge product of the gyratory crusher to the Symons disk crusher of unit 1. It also serves as a picking belt for tramp materials and is driven by a 10-hp. motor through a speed reducer and gear and pinion. The head pulley is equipped with a Ding's electromagnet. See Table 13 for operating details.

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Belt conveyor 10, Figure 2, feeds the Traylor rolls. It handles the undersize product of the grizzly ahead of the gyratory crusher, the disk crusher product and the oversize product of the "Hum-mer" screen following the rolls. It is driven through a speed reducer by a 10-hp. motor. See Table 13 for operating details.

Belt conveyor 12, Figure 2, carries the roll product to the "Hum-mer" screen and is driven through a speed reducer by a 10-hp. motor. See Table 13 for operating details.

Belt conveyor 14, Figures 2 and 3, carries the finished product of the crushing plant to the sample mill. It is driven through gear and pinion by a 50-hp. motor.

Belt conveyor 16, Figure 3, carries the oversize product of the trommel of Unit 2 to the Symons disk crusher. It is driven through gear and pinion by 10-hp. motor. See Table 13 for operating details.

Bucket elevator 18, Figure 3, elevates the discharge products of the Symons disk crushers to the trommel. It handles 150 tons of minus 2-inch material per hour, is driven by a belt from a countershaft, and consumes approximately 15 hp. See Table 13 for operating details.

Belt conveyor 21, Figure 3, is horizontal and conveys the discharge products of the 36-inch Symons disk crushers to elevator 18. It handles 50 tons of minus 1-inch material per hour and is driven by a countershaft through gear and pinion. See Table 13 for operating details.

SAMPLE MILL CONVEYING

Bucket elevator 10, Figure 4, is used to handle the various reject products of the sample cutters and discharges these products into the fine-ore storage bins. It is driven from a countershaft by gear and pinion. See Table 13 for operating details.

CONCENTRATOR CONVEYORS

Belt conveyor 2, Figure 5, is the main ore conveyor and transports the ore from the fine-ore storage bins to the head of the concentrator. It handles approximately 62.5 tons of minus 1-inch material per hour, is driven from a countershaft through gear and pinion and requires about 12 hp. for operation. See Table 13 for operating details.

Two bucket elevators, No. 6, Figure 5, elevate the roll products to the "Hum-mer screens located at the top of the concentrator. These elevators handled 62.5 tons of minus ¾-inch material per hour and are driven from a countershaft through gear and pinion. See Table 13 for operating details.

Belt conveyor 17, Figure 5, is partly inclined and partly horizontal. It receives ore intermittently from main ore conveyor 2 by means of a deflector. It discharges over a traveling tripper into a surge bin and is driven through a speed reducer. See Table 13 for operating details.

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Two pan conveyors, No. 19, Figure 6, serve as feeders from the surge bin, one under either end of the bin. These feeders discharge onto horizontal belt conveyor 20 and each has a maximum capacity of 25 tons per 24 hours. Under normal operating conditions but one of these pan conveyors is operated at a time. They are driven by chain and sprocket from a motor through a speed reducer. See Table 13 for operating details.

Belt conveyor 20, Figure 6, is a horizontal conveyor which delivers feed, received from pan conveyors, No. 19, Figure 6, to the Marcy rod mill. It is driven through shaft and bevel gears by the motor which operates the two pan conveyors, No. 19. See Table 13 for operating details.

Table 1.- Screen analyses of the products of coarse crushing plant units 1 and 2

The capacity of each unit amounts to 125 tons of ore per hour

Screen sizes	Unit 1		Unit 2	
	Individual weight, per cent	Cumulative weight, per cent	Individual weight, per cent	Cumulative weight, per cent
Plus 0.752 inch	- -	- -	4.5	4.5
Minus 0.752 inch on 0.525 inch	13.4	13.4	13.9	18.4
Minus 0.525 inch on 0.371 inch	11.9	25.3	11.8	30.2
Minus 0.371 inch on 10 mesh	38.4	63.7	29.5	59.7
Minus 10 mesh on 14 mesh	3.8	67.5	3.7	63.4
Minus 14 mesh on 35 mesh	7.3	74.8	10.7	74.1
Minus 35 mesh on 200 mesh	15.4	90.2	13.3	87.4
Minus 200 mesh	9.8	- -	12.6	- -
Totals	100.0		100.0	

Table 2.- Screen analyses of intermediate crushing and primary grinding products

Screen size	Final product of intermediate crushing, 50 to 55 tons crushed per hour		Final product of primary grinding, 50 to 55 tons of solids per hour, pulp contains 50 per cent of solids	
	Individual weights, per cent	Cumulative per cent	Individual weights, per cent	Cumulative per cent
Plus 3 mesh	4.7	4.7	- -	- -
Minus 3 mesh plus 10 mesh	65.5	70.2	- -	- -
Minus 10 mesh plus 35 mesh	13.8	84.0	25.8	25.8
Minus 35 mesh plus 48 mesh	- -	- -	10.7	36.5
Minus 48 mesh plus 65 mesh	- -	- -	11.0	47.5
Minus 65 mesh plus 100 mesh	[1]3.7	87.7	6.0	53.5
Minus 100 mesh plus 150 mesh	- -	- -	7.8	61.3
Minus 150 mesh plus 200 mesh	[2]3.1	90.8	2.8	64.1
Minus 200 mesh	9.2	- -	35.9	- -

1 Minus 35 mesh plus 100 mesh sizes. 2 Minus 100 mesh plus 200 mesh sizes

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Table 3.- Screen analyses of classifier feeds and products of regrinding department

Screen size	Dorr bowl classifier						Dorr circulating classifiers					
	Feed, 50 to 55 tons of solids per hour, pulp contains 50 per cent of solids		Sand, 25 to 30 tons of solids per hour, pulp contains 74 per cent of solids		Overflow, 25 tons of solids per hour, pulp contains 17 to 20 per cent of solids		Feed,[1] 115 to 140 tons of solids per hour, pulp contains 65 to 70 per cent of solids		Sand, 90 to 110 tons of solids per hour, pulp contains 81 per cent of solids		Overflow, 25 to 30 tons of solids per hour, pulp contains 27.8 per cent of solids	
	Individual, per cent	Cumulative, per cent	Individual, per cent	Cumulative, per cent	Individual, per cent	Cumulative, per cent	Individual, per cent	Cumulative, per cent	Individual, per cent	Cumulative, per cent	Individual, per cent	Cumulative, per cent
Plus 35 mesh	25.8	25.8	29.8	29.8	- -	- -	19.1	19.1	22.6	22.6	(2)	- -
Minus 35 mesh plus 48 mesh	10.7	36.5	20.8	50.6	1.2	1.2	20.4	39.5	22.4	45.0	11.0	11.0
Minus 48 mesh plus 65 mesh	11.0	47.5	22.6	73.2	4.2	5.4	23.8	63.3	23.8	68.8	18.6	29.6
Minus 65 mesh plus 100 mesh	6.0	53.5	7.8	81.0	5.2	10.6	10.4	73.7	13.6	82.4	10.2	39.8
Minus 100 mesh plus 150 mesh	7.8	61.3	8.0	89.0	9.7	20.3	10.2	83.9	9.4	91.8	13.4	53.2
Minus 150 mesh plus 200 mesh	2.8	64.1	2.5	91.5	5.0	25.3	2.4	86.3	2.0	93.8	5.0	58.2
Minus 200 mesh	35.9	- -	8.5	- -	74.7	- -	13.7	- -	6.2	- -	41.8	- -

1 The feed to the regrinding mills consists of the bowl classifier sand product plus approximately 2-1/2 tons per hour of tertiary flotation concentrates.

2 The excessive amount of coarse material in the circulating classifier overflow products is due to the crowded condition of the regrinding unit, which condition will be relieved by the installation of one or more grinding mills and classifiers. The ultimate aim of grinding is to produce 5 to 6 per cent plus 65-mesh material in classifier overflow products.

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Table 4.- Screen analyses of feed and products of new Marcy-mill grinding unit

Screen size	Marcy rod mill feed, 10 to 12 tons per hour containing 2.5 per cent of moisyure		Marcy mill ground product, 31 to 35 tons per hour, pulp contins 70 per cent of solids		Classifier sand product, 21 to 25 tons per hour, pulp contains 80 per cent of solids		Classifier overflow product, 10 to 12 tons per hour, pulp contains 20 per cent of solids	
	Individual weights, per cent	Cumulative, per cent	Individual weights, per cent	Cumulative, per cent	Individual weights, per cent	Cumulative, per cent	Individual weights, per cent	Cumulative, per cent
Plus 0.725 inch	2.3	2.3	- -	- -	- -	- -	- -	- -
Minus 0.725 inch plus 0.525 inch	13.7	16.0	- -	- -	- -	- -	- -	- -
Minus 0.525 inch plus 0.371 inch	11.9	27.9	- -	- -	- -	- -	- -	- -
Minus 0.371 inch plus 10 mesh	33.9	61.8	- -	- -	- -	- -	- -	- -
Minus 10 mesh plus 35 mesh	12.6	74.4	19.0	19.0	21.6	21.6	- -	- -
Minus 35 mesh plus 48 mesh	- -	- -	16.9	35.9	21.8	43.4	5.5	5.5
Minus 48 mesh plus 65 mesh	- -	- -	22.6	58.5	28.7	72.1	14.0	19.5
Minus 65 mesh plus 100 mesh	- -	- -	9.1	67.6	10.6	82.7	9.4	28.9
Minus 100 mesh plus 150 mesh	- -	- -	9.0	76.6	7.4	90.1	11.8	40.7
Minus 150 mesh plus 200 mesh	[1]14.4	88.8	2.9	79.5	3.2	93.3	7.1	47.8
Minus 200 mesh	11.2	- -	20.5	- -	6.7	- -	52.2	- -

1 Minus 35 plus 200 mesh sizes.

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Table 5.- Operating data of treatment by the Rork-Callow machines compared with treatment by the first complete installation of Forrester machines

	Rork-Callow machines combination	All Forrester machines
Period of treatments	Year 1924.	First 7 months of 1928.
Ore treated per 24 hours, tons	967	1,296
Machines used:		
Primary cells	8 Rork machines.	4 machines 15 feet long.
Secondary cells	16 Callow 2 by 6 foot machines.	4 machines 17 feet 4 inches long.
Cleaner cells	4 Callow 2 by 8 foot machines.	1 machine 10 feet long.
Air requirements:		
Total air from blowers, cubic feet of free air per minute	7,500	7,500
Air pressure at blower, pounds per square inch	4.5 to 6.0	2.5
Air pressure at cell header, pounds per square inch	----	1.37
Power for flotation, kw.h. per ton of ore:		
Primary machines	2.43	0.60
Secondary and cleaner machines	2.72	.73
Total	5.15	1.33
Labor for flotation operations, 8-hour man-shifts per day:		
Operators	3	3
Helpers	3	0
Maintenance costs per ton of flotation feed:		
Rork machines, labor and supplies	$ 0.00145	----
Callow machines		
Labor	.00408	----
Supplies	.01206	----
Forrester machines	----	To date maintenance costs are practically nil.

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Tagle 6.- Average analyses of concentrator feed and products

	Year 1929			January to May, 1930		
	Feed	Concentrates	Tailings	Feed	Concentrates	Tailings
Weight, per cent	100.0	12.4	87.6	100.00	8.63	91.37
Total copper, per cent	2.61	18.97	.28	2.37	24.43	.26
Sulphide copper, per cent	2.50	- -	.20	2.26	- -	.19
Oxide copper, per cent	.11	- -	.08	.11	- -	.07
Iron, per cent	14.3	31.4	10.2	12.6	28.6	10.6
Alumina, per cent	9.0	1.9	10.2	9.5	1.9	10.4
Sulphur, per cent	11.7	36.8	7.5	7.9	32.9	5.1
Silica, per cent	49.3	6.0	56.3	53.1	6.3	57.2
Moisture, per cent	2.5	11.9	- -	2.5	12.8	- -
Silver, ounces per ton	.35	2.22	.04	.21	1.89	.02
Gold, ounces per ton	.012	.076	Trace	.010	.079	Trace

Table 7.- Distributions of copper in concentrator feed and products, by screen sizes, for May, 1930

Screen sizes	Per cent weight		Assay, per cent copper		Per cent of total copper content	
Flotation feed:						
Plus 48 mesh	6.2	6.2	0.41	0.41	1.1	1.1
Plus 65 mesh	11.8	18.0	.61	.54	3.1	4.2
Plus 100 mesh	8.9	26.9	1.51	.86	5.7	9.9
Plus 150 mesh	12.6	39.5	2.49	1.39	13.4	23.3
Plus 200 mesh	5.2	44.7	3.06	1.36	6.8	30.1
Minus 200 mesh	55.3	- -	2.98	2.36	69.9	- -
Monthly composite	- -	- -	- -	2.53	- -	- -
Flotation concentrates:						
Plus 65 mesh	2.1	2.1	23.64	23.64	1.9	1.9
Plus 100 mesh	5.9	8.0	23.46	23.50	5.5	7.4
Plus 150 mesh	15.2	23.2	23.08	23.15	13.9	21.3
Plus 200 mesh	7.8	31.0	23.80	23.35	7.4	28.7
Minus 200 mesh	69.0	- -	25.94	25.13	71.3	- -
Monthly composite	- -	- -	- -	25.22	- -	- -
Flotation tailings:						
Plus 48 mesh	6.8	6.8	.41	.41	10.7	10.7
Plus 65 mesh	12.7	19.5	.27	.32	13.0	23.7
Plus 100 mesh	9.2	28.7	.22	.29	7.6	31.3
Plus 150 mesh	12.4	41.1	.17	.25	8.0	39.3
Plus 200 mesh	5.0	46.1	.16	.24	3.0	42.3
Minus 200 mesh	53.9	- -	.28	.26	57.7	- -
Monthly composite	- -	- -	- -	.28	- -	- -

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Table 8.- Mineralogical analyses of concentrator heads and products for December, 1928

	Mineralogical analyses, per cent			Indicated recovery of minerals, per cent	Per cent of total copper		
	Heads	Concentrates	Tailings		Heads	Concentrates	Tailings
Chalcocite	0.15	3.13	0.06	61.17	3.90	11.16	18.21
Chalcopyrite	6.46	38.10	.23	97.02	72.69	58.72	30.18
Bornite	.87	9.47	.06	93.69	17.95	26.76	14.44
Covellite	.01	.39	Trace	- -	.23	1.15	- -
Cuprite	.18	.56	.11	48.40	5.23	2.21	37.17
Native copper	Trace	Trace	Trace	- -	- -	- -	- -
Pyrite	10.81	35.38	8.23	31.10	- -	- -	- -
Gangue	81.52	12.97	91.31	2.09	- -	- -	- -
Totals	100.00	100.00	100.00	- -	100.00	100.00	100.00

Table 9.- Screen analyses of feed and products of classifiers operating at tailings dam

Screen sizes	Feed: Flotation tailings pulp containing 15.6 per cent of solids and from 1,350 to 1,400 tons of solids per day		Sand: This product is used to build dam and contains 72 per cent of solids and from 660 to 685 tons of solids per day		Overflow: This product goes to pond behind dam and contains 13.1 per cent of solids and from 690 to 715 tons of solids per day	
	Individual weights, per cent	Cumulative, per cent	Individual weights, per cent	Cumulative, per cent	Individual weights, per cent	Cumulative, per cent
Plus 48 mesh	10.0	10.0	21.1	21.1	0.3	0.3
Plus 65 mesh	14.8	24.8	27.8	48.9	1.7	2.0
Plys 100 mesh	9.2	34.0	16.3	65.2	3.1	5.1
Plus 150 mesh	10.1	44.1	13.3	78.5	6.3	11.4
Plus 200 mesh	7.6	51.7	7.6	86.1	7.2	18.6
Minus 200 mesh	48.3	- -	13.9	- -	81.4	- -

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Table 10.- Metallurgical data for the year 1929 and first five months of 1930

	Year 1929	First five months of 1930
Total ore treated, tons	454,019	213,431
Days operated	339.33	140.83
Operating time per day, hours	24	24
Average ore treated per operating day, tons	1,338	1.515
Total concentrates produced, tons	56,062	18,423
Concentrates produced per operating day, tons	165.5	130.8
Indicated recovery of copper, per cent	90.64	89.99
Actual recovery of copper, per cent	89.91	89.07
Loss unaccounted for, per cent	.73	.92
Recovery of sulphide copper, per cent	93.84	93.33
Recovery of oxide copper, per cent	39.90	41.90
Ratio of concentration, tons into one	8.10	12.36
Heads assays:		
Sulphide copper, per cent	2.501	2.26
Acid-soluble copper, per cent	.109	.11
Total copper, per cent	2.61	2.37
Concentrates assays:		
Copper, per cent	18.97	24.43
Insoluble, per cent	7.86	6.90
Iron, per cent	31.36	28.75
Sulphur, per cent	36.75	33.15
Tailings assays:		
Sulphide copper, per cent	.203	.19
Acid-soluble copper, per cent	.076	.07
Total copper, per cent	.279	.26
Plus 65-mesh material, per cent	19.4	21.1
Net water consumption per ton of ore, gallons	1,105	1,025
Consumption of balls and rods per ton of ore, pounds	2.48	2.41
Consumption of liners per ton of ore, pounds	.29	.21

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Table 11.- Summary of direct concentrator costs for the year 1929

Ore treated = 454,019 tons. Concentrates produced = 56,062 tons

	Cost per ton of mill feed
Crushing and conveying	$ 0.086
Secondary crushing	.064
Primary grinding	.094
Secondary grinding	.093
Flotation	.140
Concentrates handling	.042
Tailings disposal	.025
Sampling and assaying	.044
Mill water	.029
Miscellaneous	.082
Total direct costs	$ 0.699
Cost by elements:	
Total labor	$ 0.220
Supply-department issues	.089
Flotation reagents	.069
Grinding materials	.063
Power	.173
Water	.029
Heat	.004
Sample mill and laboratory	.034
Miscellaneous	.018
Year 1929, total direct costs	$ 0.699
Year 1928, total direct costs	,764

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Table 12.- Summaries of costs in units of labor, power and supplies for the year 1929

Ore concentrated = 454,019 tons. Concentrates produced = 56,062 tons.

	Operating	Repairs	Total
Labor, tons of ore treated per man-shift	36.7	92.3	26.4

	Per ton of ore concentrated
Distribution of labor:	
Primary crushing and conveying	$ 0.43
Secondary crushing and conveying	.019
Primary grinding	.019
Secondary grinding	.018
Flotation	.023
Concentrates handling	.020
Tailings disposal	.015
Sampling and assaying	.006
Concentrator water	.001
Concentrator department expense	.056
Total	$ 0.220

	Kw. h. per ton of ore treated	Per cent of total power
Distribution of power:		
Primary crushing and conveying	1.38	8.65
Secondary crushing and conveying	1.07	6.70
Sample mill	.12	.75
Primary grinding	3.88	24.31
Secondary grinding	5.01	31.39
Flotation	1.55	9.71
Feed and middlings pumps	.95	5.95
Milk of lime plant	.17	1.07
Concentrates handling	.78	4.89
Tailings disposal	.74	4.64
Lights and shops	.31	1.94
Total	15.96	100.00

	Pounds per ton of ore treated
Reagents and miscellaneous supplies:	
Reconstructed oil	0.088
Pine oil	.193
Potassium ethyl xanthate	.086
Experimental	.014
Lime	9.005
Primary crushing steel	.09
Secondary crushing steel	.11
Balls and rods	2.48
Liners	.29
Total steel	2.97

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Table 13.- Data for belt conveyors and elevators

Conveyor number	2	3	7	10	12	14	16	18	21	10	2	6	17	19	20
Figure number	2 and 3	2 and 3	2	2	2	2 and 3	3	3	3	4	5	5	6	6	6
Kind of conveyor	Pan	Belt	Belt	Belt	Belt	Belt	Belt	Elevator	Belt	Elevator	Belt	2 Elevators	Belt	2 pan conveyors	Belt
Length, feet	-	214	102	157	120	583	109	116	49	59	310	130	129	-	45
Width, inches	48	36	36	24	24	24	36	20	18	12	18	20	18	-	18
Thickness, ply	-	6	5	5	5	6	6	10	5	6	6	8	5	-	5
Thickness of top cover, inches	-	1/4	3/16	3/16	3/16	3/16	3/16	1/8	3/16	1/8	1/8	3/64	1/8	-	1/8
Thickness of bottom cover, inches	-	1/16	1/32	1/32	1/32	3/64	1/32	1/8	1/32	1/8	3/64	1/8	1/32	-	1/32
Speed, feet per minute	8	103	50	100	100	327	50	325	200	275	127	430	195	8	38
Diameter of head pulley, inches	-	30	30	24	24	48	30	42	24	42	30	42	20	-	-
Diameter of tail pulley, inches	-	24	20	24	24	24	24	30	18	30	20	30	18	-	-
Slope	-	16°	20°	20°	20°	18°	18°	Vertical	Horizontal	Vertical	13°	Vertical	20° for 20 feet	- -	Horizontal
Vertical lift, feet	-	-	-	-	-	88	16	52	-	25	-	60	-	Maximum ¼-inch	Maximum ¼-inch
Size of material handled	Maximum 16-inch	maximum 16-inch	Minus 4-inch	Minus 1½ inch	Minus 1-inch	Minus 1-inch	Maximum 4-inch	Maximum 2-inch	Maximum 1-inch	-	-	-	-	4 by 24	-
Size of pans or buckets inches	8 by 48	-	-	-	-	-	-	-	-	-	-	-	See text		
Horsepower of driving motor	-	-	10	10	10	50	10								
Manner of driving	See text														

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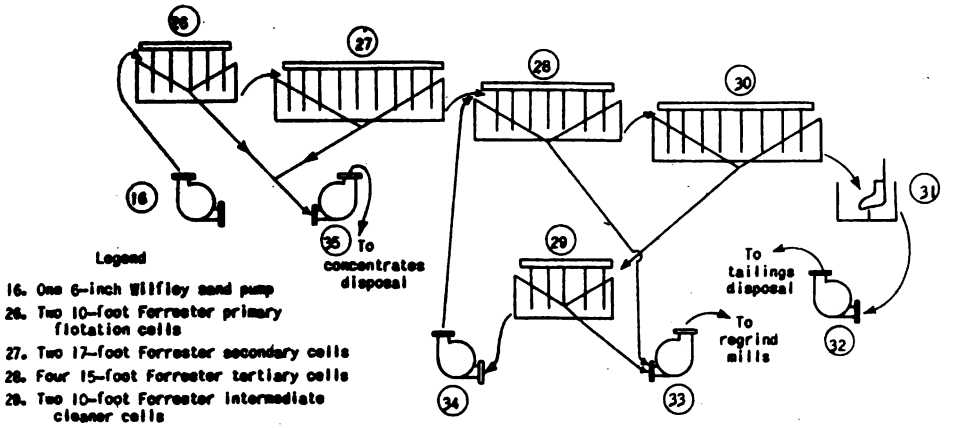
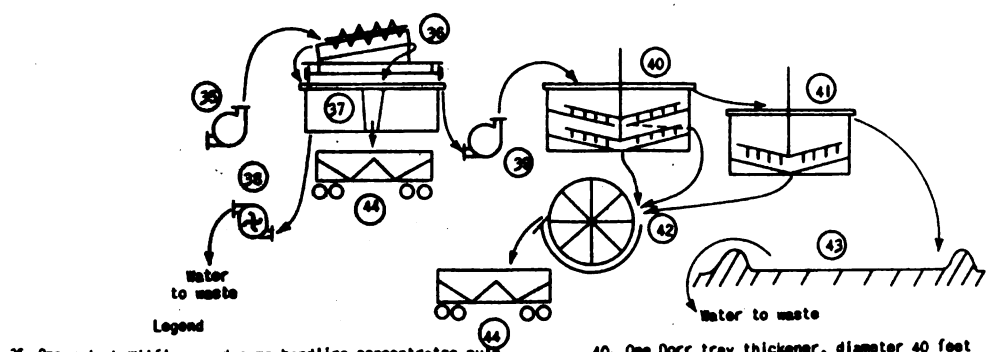


Legend

16. One 6-inch Wilfley sand pump
26. Two 10-foot Forrester primary flotation cells
27. Two 17-foot Forrester secondary cells
28. Four 15-foot Forrester tertiary cells
29. Two 10-foot Forrester intermediate cleaner cells
30. Four 17-foot Forrester scavenger cells
31. Two automatic tailings samplers
32. One 6-inch Wilfley sand pump for tailings
33. One 3-inch Wilfley sand pump for concentrates
34. One 4-inch Wilfley sand pump for middlings
35. One 4-inch Wilfley sand pump for concentrates

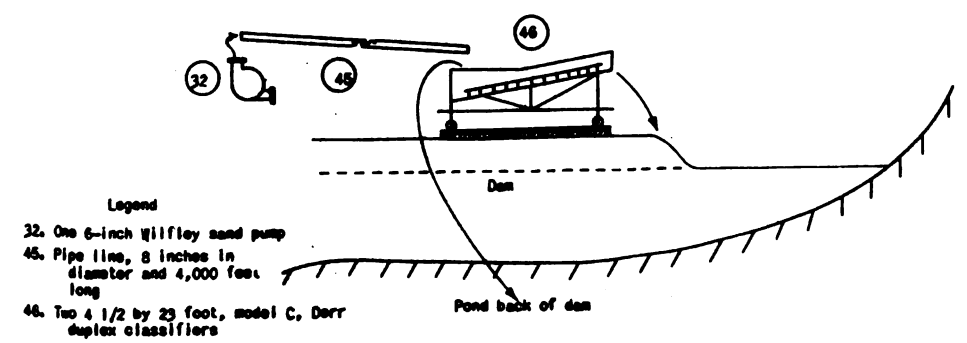
Figure 7.- Flow sheet of flotation treatment



Legend

35. One 4-inch Wilfley sand pump handling concentrates pulp
36. One 36-inch Akins classifier
37. Nine cylindrical concrete storage bins, 24 feet in diameter and 8 feet high, operated with filter bottoms
38. One No. 2 Nash Hytor rotary vacuum pump
39. One 4-inch Wilfley sand pump
40. One Dorr tray thickener, diameter 40 feet
41. One ordinary Dorr thickener, diameter 40 feet
42. Two Oliver filters, one 14 feet in diameter by 14-foot face and the other 8 feet in diameter by 12-foot face
43. Settling ponds
44. Railroad cars

Figure 8.- Flow sheet for dewatering and handling concentrates



Legend

32. One 6-inch Wilfley sand pump
45. Pipe line, 8 inches in diameter and 4,000 feet long
46. Two 4 1/2 by 23 foot, model C, Dorr duplex classifiers

Figure 9.- Flow sheet of tailings disposal

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